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MINING AND MILLING PRACTICES AT SMALL GOLD MINES

BY

E. D. GARDNER AND C. H. JOHNSON

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MINING AND MILLING PRACTICES AT SMALL GOLD MINES[1]

By E. D. Gardner[2] and C. H. Johnson[3]

INTRODUCTION

This paper discusses small-scale lode gold mining and milling. It is abstracted from a bulletin being prepared by the United States Bureau of Mines entitled "Equipping, Developing, and Operating Small Gold Mines."

Gold is widely distributed in the mountainous regions of the United States; relatively few deposits, however, can be profitably worked on a large scale. Because of the high intrinsic value of gold, ores containing this metal may be very rich; hence it is possible to mine profitably smaller bodies of gold ore than of the more common metals.

During periods of depression when the prices of base metals are low, increased interest is shown in gold mining. Moreover the increased price of gold which went into effect in 1933 greatly stimulated gold mining. From 1931 to 1934 many individuals and small groups were engaged in the exploitation of gold deposits where the ore occurred in narrow veins or as small bunches. In most cases current day's wages were not made by the workmen; in other cases, however, fair returns were obtained. During this period many old gold mines also were reopened and new gold properties developed by organized companies. Only part of the operations by companies were successful. Many of the unsuccessful companies were inadequately financed; others failed principally because of inexperienced or poor management.

GEOLOGY

Oreshoots

A careful study of the conditions pertaining to the occurrence of ore is of assistance in the search for oreshoots and in planning development work in any mine.

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6800."

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Usually the oreshoots in gold mines (that is, the portions of veins that are sufficiently rich in gold to permit profitable mining) are distinguished by the presence of certain minerals or by their relationship to one or more of many geological features. They may occur in a certain country-rock formation or only where cross-fractures intersect the vein or mineral zone. Every effort should be made to discover such "controls" or "guides" if any exist. Continual assaying of samples of distinct types of vein material and wall rock, with a careful study of the appearance and structure of the ground at the point sampled, usually yields valuable information. For instance, it may be shown that the presence of pyrite, or perhaps of amythest-colored quartz, indicates good values; again, the ore may be richest where the vein material is strongly fractured or where it contains many vugs or cavities. In some instances, swells in the walls are favorable points for ore deposition; elsewhere the gold content may be highest where the vein is narrow. Possibly the vein may contain ore only near certain dikes or may "make ore" only in one of several different flow rocks. Such discoveries increase the chance of finding new ore and help to avoid misdirected effort. If the ore is free milling, the samples may be tested quickly and cheaply by panning. This form of testing, however, should be checked frequently by assaying.

Although thin veins are sometimes extensive, as in some California mines, the general rule is that in narrow veins the oreshoots are both small and irregular in form. The first principle to be followed in developing narrow gold veins, at least until the oreshoots have been proved to be of considerable extent and regularity, is to follow the ore. It is noteworthy that even in old, productive mines where ore has been found more or less continuously for hundreds of feet from the outcrop, winzes are usually sunk on the ore to prove its continuance before the deepening of a main shaft to the next lower level is started. The most common mistake made by inexperienced operators is to run costly long drifts or, worse, crosscuts to prospect narrow veins at depth when it would have been possible to gain the same information much cheaper by sinking on known ore. Long crosscuts are seldom justified without definite knowledge that the ore extends downward.

Faulting

Faults may be premineral, postmineral, or both. If a fault that cuts through an oreshoot is older than the ore, it often appears to have acted as a barrier to the ore-bearing solutions with the result that there is no ore on the farther side of the fault. This is common when a premineral fault contains much gouge. If, however, the fault that cuts off an oreshoot took place after the ore was deposited, there is reason to hunt for the displaced ore beyond the fault. The fault is then studied to determine, if possible, the direction and distance of the movement. The evidence pertaining to faulting questions is seldom clear-cut, yet exploratory work based on geological deductions is better than pure guesswork.

When a fault is encountered underground that cuts off the ore and displaces it so far that a round or two in either direction fails to disclose its continuation, the surface is first examined carefully to see if the vein outcrop is offset by a fault that may be the same as the one found underground. If such a fault is found the direction to seek the vein underground should be apparent. Even if the fault does not cut the vein outcrop it may have displaced dikes, sedimentary beds, or contacts in such a way as to give an idea of the direction and amount of displacement.

A study of the fault breccia and fault surfaces sometimes gives a clue to the direction of throw. If crushed fragments of vein material, especially of minerals found in the oreshoots, are found in the fault gouge, the fault is presumed to be later than the ore; the trail of crushed ore fragments indicates the direction of movement. Rock fragments in the fault zone sometimes leave tracks on the fault surface. If such a fragment can be found still embedded at the end of its furrow, it shows which way the opposite wall has moved.

Faults are not simple, relatively up-and-down movements of the two sides; usually the movements have been in oblique directions. This can be seen from the slickensides on the fault surfaces underground. Furthermore, faults seldom consist of a single clean break. Often the vein will be offset by numerous minor faults on either side of the main break. These minor faults generally show the same relative direction of displacement as the main one, hence may show which way to turn. If such minor faults are seen to exist in the country rock yet do not offset or cut the vein, it is assumed that they and the large fault to which they appear to be related are older than the ore.

If faulting is at all complex or the mine workings are extensive, a survey will be needed to show the relationship of underground and surface features or of faults and veins cut on different levels. A detailed discussion of faults is to be found in most textbooks on geology.

DEVELOPMENT

The manner of developing a small lode mine depends mainly upon the mineral showing and the financial resources of the operators. The mineral showing in the vast majority of prospects being worked is not sufficient to justify expensive surface plants. Moreover, many of the operators do not have the resources to buy such plants even where it would be economical to do so. Many ingenious and inexpensive devices for hoisting are noted in the field. Old automobile engines which are obtained for little cash outlay are used for power in many places.

When a flow of water is encountered in working from a shaft, power pumps are necessary. Water sometimes induces the operators to drive adits to give the necessary depth.

The first work done on narrow veins generally consists of opencuts. If the relief is sufficient the vein is then followed by a drift. Shafts are sunk for deeper development and raises run to explore or mine the vein above drifts. Crosscutting is seldom necessary unless more than one vein is worked or a vein is reached through a crosscut from a shaft. Most of the exploratory and development footage at small gold mines consists of drifting. In the early stages of the work, at least, these openings are driven by hand. The drifts generally are about 6 feet high and 4 1/2 feet at the widest part; they are narrower at the top, and the back is usually arched. Drifts are run as small as practicable to save in first cost and to reduce the need of timbering; the minimum size generally is governed by the size of a tramming car.

Hand-drilled rounds which are seldom over 2 or 3 feet deep can be broken readily in these narrow workings. However, where machine rounds are drilled, usually it is necessary to widen the drift to permit a longer round to be pulled.

A common fault of exploratory drifts is that the grade is so steep and uneven that tramming is difficult. A straight edge made of a 1- by 4- by 12-foot board and a carpenter's level generally can be used to good advantage for indicating the grade. A block of the thickness of the desired slope is used on the outward end of the straight edge. A grade of 3/4 inch to 1 1/2 inches to 12 feet will make hand tramming easy and give good drainage.

Attention given to the position of a development drift on a narrow vein with reference to the vein will be of value when ore above the drift is stoped. If the vein is quite steep the drift may be run equally well with the vein on either side or in the middle. If the vein dips 60° or less it is more convenient to have the drift in such a position that the stope floor will intersect the side of the drift high enough to install chutes for loading a car or a bucket. Figure 1 shows three drift positions; in flat veins the drift is run as far under the vein as possible without losing sight of it entirely.

There is usually no difference in the size or shape of drifts and crosscuts. Crosscuts commonly require less timber than drifts because country rock away from the ore zone is usually less fractured and altered and is therefore stronger. When drifts require support it is sometimes possible to utilize single post and cap sets in which one end of the cap is hitched into the footwall. Heavy timbering may be required near old workings but usually is not necessary in new work.

Shaft sinking is the most costly form of development work and requires the largest initial expenditure; the footage of shafts, however, is relatively small compared to that of drifts. Inclined shafts are commonly sunk on the vein at small gold mines; this practice permits following the ore down and eliminates the necessity of crosscuts to reach the vein, as required when a vertical shaft is sunk in the wall rock. Timbering in an inclined shaft is simpler and cheaper than in a vertical shaft. The timbering in an inclined shaft is shown in figure 2,A, and in a vertical shaft in figure 2,B. Vertical shafts must be timbered even if the ground does not require the support in order to provide a safe manway and to hold guides. For all depths over 75 feet guides should be used.

In a small mine the inconvenience and limited hoisting capacity of an inclined shaft usually are not serious enough to justify sinking a vertical shaft in the country rock. Furthermore, stoping in a narrow vein in the vicinity of a shaft is not likely to cause the shaft to cave as it frequently does in wide deposits.

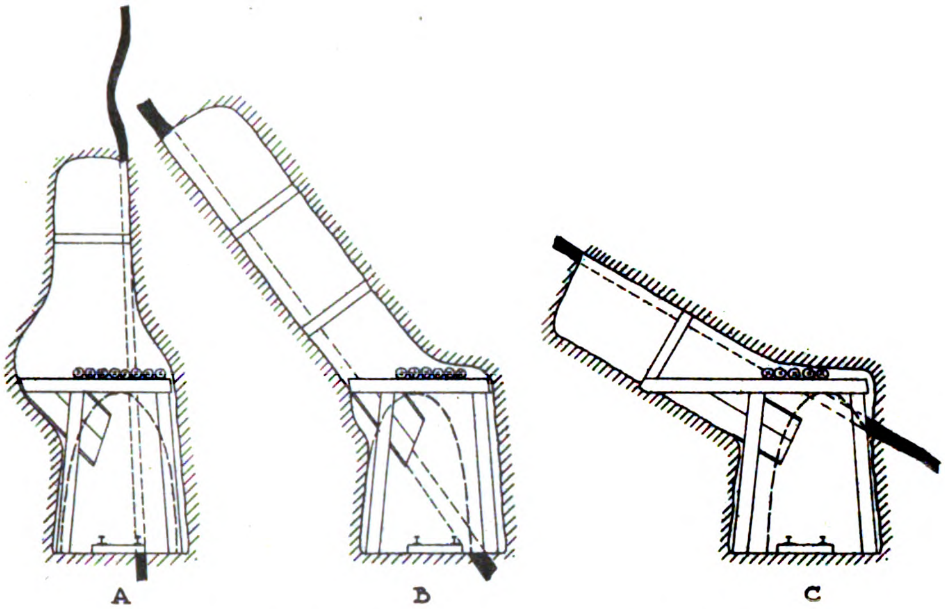

Fig. 1.—Various positions of drifts with relation to veins and stopes.

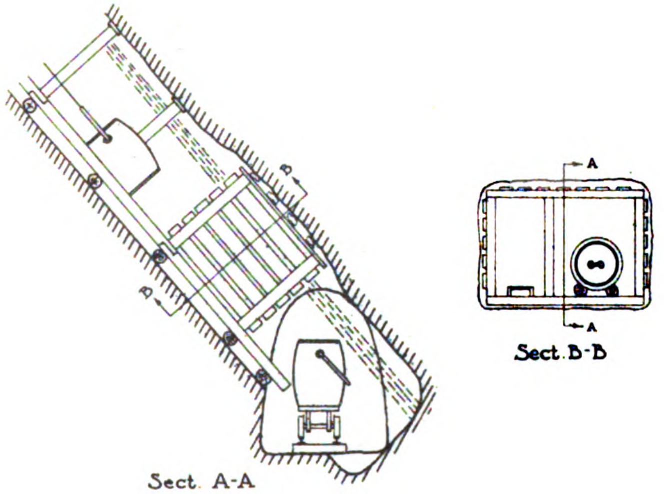

Fig. 2A.—Inclined shaft for hoisting with bucket.

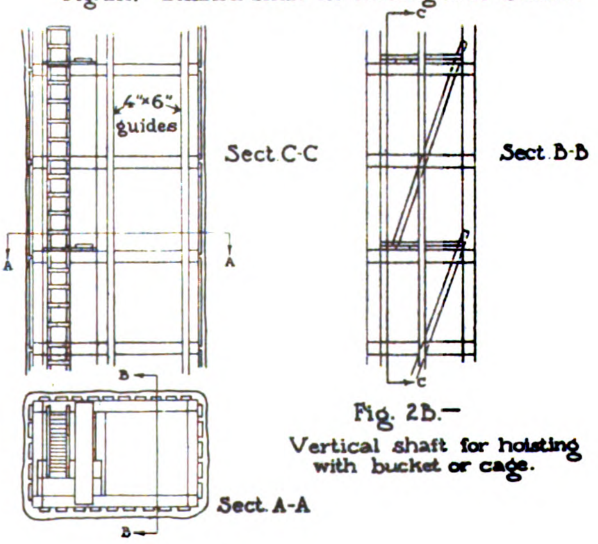

Fig. 2B.—
Vertical shaft for hoisting
with bucket or cage.

At the start of operations sinking usually is done by hand; hand windlasses may be used up to 100 feet or even deeper. For depths over 50 feet, however, power hoists are more economical.

Raising is of less relative importance in thin veins than in wide deposits. Some raising usually is necessary, however, for access to oreshoots that terminate above a given level, for ventilation and entrance to stopes as they approach levels above, and for emergency exits to the surface. Raises usually are run about 3 1/2 or 4 by 7 feet in section and timbered with stulls. If the dip is over 45° the raises generally contain two compartments, one for a manway and the other for an ore chute.

In mining narrow veins a relatively small tonnage of ore is made available by a given amount of development; in other words, development cost per ton of ore is high. The following table show the tons of ore developed per foot of drift in narrow veins with 50-foot lifts.

Average thickness of ore, inches	Tons of ore per foot of drift	Average thickness of ore, inches	Tons of ore per foot of drift
3	1.0	24	8.3
6	2.1	36	12.5
12	4.2		

The cost per ton, for example, in a 6-inch vein would be the cost per foot of drifting divided by 2.1. If drifting cost $5 per foot, the development charge for this item alone would be $2.40 per ton, to which must be added the usually greater cost of drifting in barren portions of the vein, as well as crosscutting, raising, and shaft sinking.

The direct cost of drifting by hand drilling is about the same as that with power drills except in very hard rock. The progress, however, with an equal number of men working is only about one third as fast when drilling by hand as when air drills are used; thus, when there is any overhead the total cost of hand drilling is higher. Most work done by hand is near the surface where the rock has been more or less altered and is therefore easier to break.

Drifting costs as low as $1 per foot have been obtained in running short adits by hand, starting at the surface. In hard rock the cost may be as high as $20. The average direct cost of drifting in all types of rock by hand is about $5 per foot. Crosscutting costs are usually higher. Progress per man-shift in drifts is usually about 1 foot. The maximum and minimum progress noted by the authors was 4 feet per man-shift in picking ground and 4 inches in tough limestone--both at prospects in Montana.

The total cost per foot of development work under the same natural conditions varies widely. Men working for themselves or on a contract may work

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a full shift underground and then sharpen their steel and do other miscellaneous jobs after hours. Moreover, in running relatively short adits with power drills they may get along without a compressor operator. Men working for day's pay, however, cannot be expected to put in overtime or do these extra jobs.

Where work is done on company account, supervision must be provided and other overhead expenses incurred. The effect of overhead on costs is shown by the following example. An adit run by the two owners of a claim cost a little under $5 per foot; this included the prorated cost of second-hand blacksmith tools, track, and a 3/4-ton car. Later a company was formed, a compressor was installed, and two headings were extended. The direct cost still remained close to $5; the total cost per foot, however, was increased by the salary of a superintendent and office expense in a distant city, making a total cost of $8 per foot. The property was then taken over by an exploration company, more equipment installed, and the work continued. The total cost per foot including engineering and central office expense was over $15 per foot for the work done.

Development drifts run from shafts are usually more costly than adits. Pumping, hoisting, and top expenses may multiply the direct drifting cost several fold. As an example, the cost per foot of extending a single drift from a 200-foot interior shaft was over $40 in a Colorado gold mine where 300 gallons of water was pumped per minute.

In a producing mine the total cost of drifting done before ore was discovered on two deep levels, as figured for income-tax purposes, was $20 and $23 per foot, respectively. The direct costs were $9 and $9.60.

The cost of prospect shafts ranges from $7 to $60, depending upon the natural conditions and the size and depth of the shaft. Table 1 shows a range in the cost of prospecting shafts.

MINING

Veins of gold ore are worked where the gold may be contained in a single streak or seam down to less than an inch wide. Where narrow streaks are stoped, waste adjoining the ore must be broken to provide working room. The minimum working distance required varies in different localities. In western metal mines being worked on company account it is customary to break enough ground (5 to 7 feet) that men may stand erect or nearly so in flat deposits or that they may work without rubbing their arms in steeply inclined veins (3 to 4 feet). Lessees, however, may work while kneeling down or lying on their sides in flat, narrow veins. The minimum distance noted by the author was 18 inches in a vein that dipped 28° at the Herron and Laster lease at Superior, Ariz.[4] Mexican contract miners in the Arizona asbestos mines work in flat stopes 16 to 18 inches high; American metal miners, however, rarely follow these practices. The breaking of waste to provide working room is called "stripping."

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[4] Gardner, E. D., Mining Methods and Costs at Herron and Laster Lease, Superior, Ariz.: Inf. Circ. 6799, Bureau of Mines, in press.

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TABLE 1. - Cost per foot of prospecting shafts

Mine	Location	Shaft			Manner of drilling	Kind of hoist	Number of men	Cost per foot
		Section feet	Depth, feet	Inclination, degrees				
Gold Coin ...	Silver Lake, Mont.	4 by 5	0 to 140	70	Hand	Horse whim.	3	1/$16.00
Rabbit	Baker, Oreg.	4 1/2 by 7	75 to 140	90	Air	Gasoline	4	2/ 40.00
Bullion Dome.	Prairie City, Oreg.	4 by 6	0 to 32	65	Hand	Hand	2	7.70
Gold Hill....	Gold Hill, Utah	6 by 8	150 to 316	80	Air	Gasoline	3	3/ 15.50
Gold Queen ..	Beaver, Utah.	3 1/2 by 7	0 to 60	90	Hand	Hand	3	4/ 7.01
Mossback Extension	Oatman, Ariz.	7 by 10	65 to 104	90	Air	Gasoline	4	5/ 14.35
Maxwell shaft no. 1	Prescott, Ariz.	4 by 6	0 to 40	90	do	Hand	1	6/ 16.00
Maxwell shaft no. 2	do	5 by 9	0 to 16	90	do	do	1	7/ 21.00
Comstock-Dexter	Bagdad, Ariz.	6 1/2 by 12	0 to 95	90	do	do	6	8/ 32.03

1/ One half superintendent's time charged.
2/ On day's pay; no supervision.
3/ Direct cost only; no overhead charged.
4/ Total cost; no overhead.
5/ All costs included, but wages were only $1.50 per day and timber was salvaged material.
6/ Including $15 for labor at $4 per shift.
7/ Including $20 for labor at $4 per shift.
8/ Including one half the pay of a blacksmith, but excluding supervision and overhead expense.

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Stopes ordinarily can be much narrower in steeply dipping deposits than in flat beds. Usually enough ground is broken in steep veins that a man can stand and work facing the strike of the deposit. A small man, of course, can work in a narrower place than a large one. The minimum width in which a man can work efficiently is about 24 inches. The senior author, however, has seen ancient stopes in Mexico that had a maximum width of 15 inches between walls. Mexican lessees in the Southwest sometimes carry stopes about 18 inches wide. The usual minimum range in producing mines where steeply pitching, narrow seams are mined is 2 1/2 to 4 feet.

The minimum thickness carried, within a range of a few inches, will depend upon the width of the vein. That is, a man will work in an uncomfortably narrow place before he will break any wall rock. If one of the walls, however, must be broken, enough rock usually is blasted to allow plenty of working room. In hard rock a wider opening is necessary than in soft material in order that the ground may be efficiently broken by the explosives.

Another consideration in stoping is the volume of waste broken in getting out the ore. The volume of rock increases about 50 percent after being broken from the solid. It is desirable, therefore, in mining narrow seams of ore to break only enough waste that the expansion or "swell" will just fill the space occupied by the ore. As an illustration, where the ore is 1 foot thick 2 feet of waste after being broken would just fill the stope, disregarding the space taken up by chutes and manways. Usually, however, chutes and manways occupy a considerable part of narrow stopes.

Generally wider stopes are taken up where machine drills are used than where the ore is mined by hand under similar conditions. Where the ore is soft it is usually picked out or drilled and blasted lightly. After the ore is removed from the stope the waste on one side is drilled and blasted and used for filling in the stope. All waste in excess of that needed for filling must be removed. Where the ore is harder than the waste or is "frozen" on a wall the waste usually is broken first. In flat veins or beds working room is provided by removing waste from over the ore wherever practicable. After stripping the ore may be either drilled and blasted or picked down.

Clean mining is very desirable in small-scale mining. That is, the ore should be kept separate from the waste or vein filling which is too low grade to pay for mining by itself. Frequently in gold deposits the gold occurs in narrow streaks in the vein. Occasionally, however, the streaks may be so irregular that they cannot be broken separately and considerable waste must be included with the ore. In general, wherever practicable, all material not paying its own way should be left behind in the stopes. Under some conditions the saving in mining costs per ton made possible by mining a wider orebody may make it more profitable to take out all material between walls. However, in small-scale mining as discussed in this paper this condition seldom arises.

Cleaner ore can be broken and narrower seams mined separately when hand drilling is followed. Air drills may be used for development work, but the stoping should be done by hand to produce a cleaner product. For instance, at the Palmer mine, Gold Hill, Utah, the ore is mined by hand where 18 inches or less thick, and where over this width drilling is done with air drills. At the mine of the St. Joseph Lead Co., at Atlanta, Idaho, no hand drilling is done; here the minimum width of stopes is 4 feet. Where the ore is less than this thickness it is first broken and removed, then enough waste is shot down to produce working room and to provide filling. Where the ore is less than 18 inches thick the portion of the vein containing the ore is broken to this width and sent to the mill, then 2 1/2 feet of the vein is shot down.

Where stripping is necessary the stoping cost per ton of ore increased as its thickness decreases. The following table shows the tonnage of waste broken to each ton of ore in streaks of different thickness in a 30-inch stope and the tonnage of waste that must be removed with each ton of ore. It is assumed that the ore and waste have the same specific gravity.

Width of ore	Waste broken per ton of ore removed, tons	Waste removed per ton of ore removed, tons
1 .... inch	29	14
3 .... do	9	4
6 .... do	4	1
9 .... do	2.33	0.17
1 .... foot	1.5	0
1.5 .... do	.17	0
2 .... do	.25	0
2.5 .... do	0	0

Stoping Methods

Cut-and-fill, open-stoping, or shrinkage methods are used in most small-scale gold mines. On the whole simpler practices are followed than in large deposits. Where stripping or sorting is necessary at the face a cut-and-fill method generally is adopted. If all the material broken is removed, open stopes usually are preferred. Shrinkage stoping may be used in steeply dipping veins where the ore is 3 feet or more thick and the walls require support afforded by the broken ore. In very heavy ground a modified form of square-setting may be followed.

Small or erratic oreshoots are generally mined out as found; no fixed routine or planned procedure is practicable. Where the ore is fairly extensive, however, it is attacked systematically. Where small tonnages are handled in steeply dipping veins the expense of ore chutes may not be justified and the ore may be dropped to the drift below where it is shoveled up by hand. Furthermore, if the ore is hoisted in buckets, chutes may not be practicable on account of the easy blocking of the small gates required for loading the buckets. In most cases, however, chutes are used, especially in cut-and-fill stopes.

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The ventilation of small stopes may present a problem. Wherever the size of the orebody permits, a raise is usually run to the level above before stoping begins. In open stopes from a drift the natural circulation of air currents may suffice. Where such stopes open from a raise, either a second opening or mechanical ventilation is required. Small stopes can be ventilated adequately through tubing by means of a blower where either compressed air or electric power is available. Wherever practicable, however, a second opening should be provided in all stopes. The filling in cut-and-fill stopes and the ore in shrinkage stopes prevent the free circulation of air from below. Natural ventilation in cool mines may be sufficient if two or more openings are carried up from the level. Where drilling is done by power drills the stopes may be ventilated by compressed-air jets, but this is an expensive way of supplying air.

Cut-and-Fill Stoping

As mentioned above, the cut-and-fill method of stoping is used in most cases in steeply dipping veins where it is necessary to strip the ore or to do much sorting at the face. Back filling the waste provides a cheap method for its disposal. Stulls with headboards may be used for support in addition to the waste fill if the ground is heavy.

Stope rounds usually are drilled vertically; the first round on a cut is drilled as in starting a raise. Subsequent rounds generally consist of as many holes as a man can drill and load in a shift; this ranges from 3 to 7 holes 2 1/2 to 3 1/2 feet deep when drilling by hand to 20 or more 5- to 6-foot holes when air drills are used. Drill holes range from 1 foot apart in narrow streaks to 4 or 5 feet in wide stopes. Where sorting is necessary horizontal rounds may be drilled. This system of breaking limits the number of working faces in a stope and may reduce the tons broken per man-shift, but these advantages may be more than compensated by the better opportunity afforded to break ore and waste separately and thus keep the ore clean.

Stoping by this method in a small orebody in a narrow, steeply dipping vein is shown in figure 3. Shoveling is difficult in narrow stopes on account of the restricted room; a free swing of the shovel is not possible. Therefore, chutes are placed closer together in narrow stopes than in wider ones. The spacing of chutes ranges from about 20 to 50 feet, depending upon local conditions. In some stopes wheelbarrows are used and the raises spaced relatively far apart. In oreshoots up to 30 or 40 feet long only one chute and manway may be used. In small-scale mining the ore usually is broken into small fragments, hence chutes with small cross section can be used successfully unless the ore has a tendency to be sticky. The cross section of chutes ranges from 2 1/2 by 2 1/2 to 4 1/2 by 4 1/2 feet inside measurement. In stopes under 4 1/2 feet wide one dimension of the chute is usually the distance between walls.

Chutes may be cribbed (usually with 4- by 6-inch or lighter timber) or may consist of plank lining inside of sets or on stulls placed between walls. In ground that stands well, stulls for holding the chute lining may be placed in hitches in the walls about 5 feet apart vertically. In ground requiring support, sets would be used instead of stulls or cribbing.

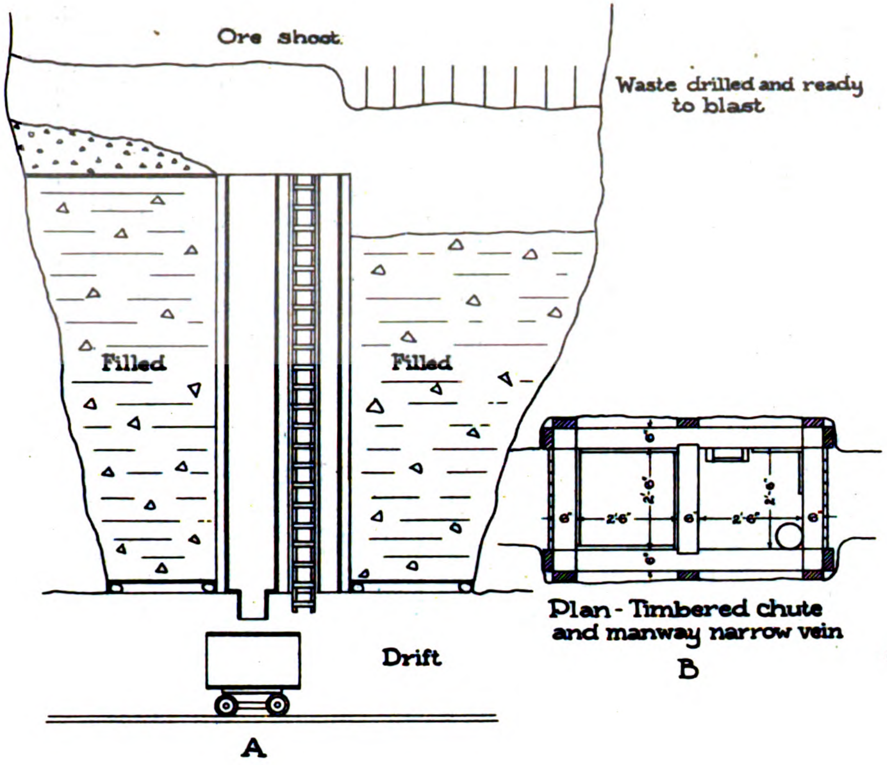
Ore shoot.
Waste drilled and ready to blast
Filled
Filled
Drift
A
6"
2'-6"
6"
2'-6"
6"
6"
6"
2'-6"
2'-6"
Plan - Timbered chute and manway narrow vein
B

Fig.3.—Vertical section, cut-and-fill stope, in small ore body.

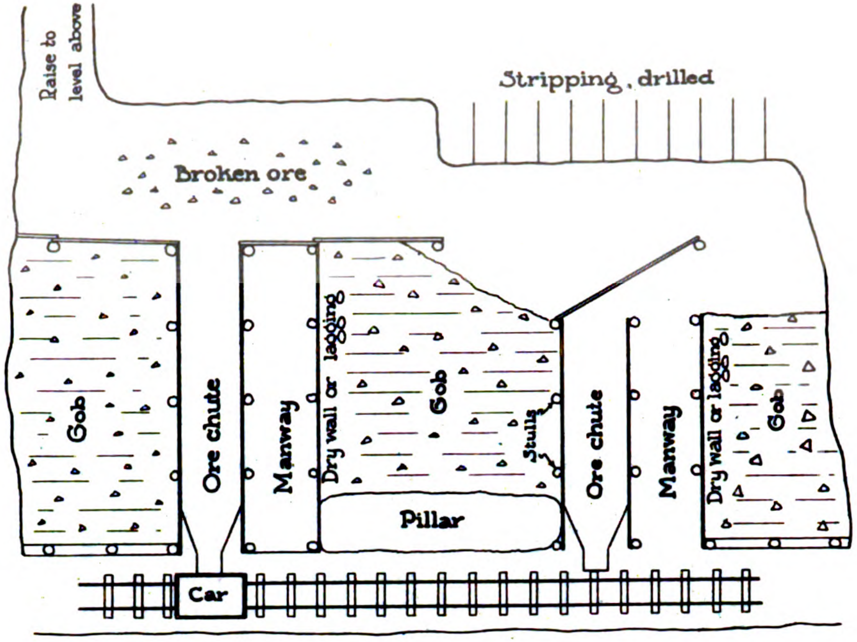

Fig.4.— Representative method of stoping relatively flat, narrow veins.

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The minimum section for a manway is 2 1/2 by 2 1/2 feet. Larger manways than this, however, are usually built. Safety rules in most States require that ladders in manways be staggered, and to do this one dimension of the manway should be at least 4 feet.

Where insufficient waste is broken in stoping to supply the required filling, one of the walls may be slabbed, waste raises run, or filling brought into the stope through raises from the level above.[5]

Care is necessary in breaking high-grade gold ore that an appreciable amount of the gold is not lost. As the grade of ore increases, greater refinements in handling are justified. The fines, which usually are very much richer in gold than the rest of the ore, are the most easily lost. In cut-and-fill stopes a medium-grade ore probably would be broken down on a plank floor in which the chinks were calked with sacking or some similar material and thence shoveled into chutes. In some places canvas is laid under the flooring. Unusually rich ore is picked down into the boxes, or if blasting is necessary it is shot lightly and caught on canvas; it is then sacked in the stope. Rich ore may be handled in the same manner with any method of stoping. Frequently it is found that enough gold gets into waste fills to justify pulling this material later for mill ore. Frequently old fills are also screened to recover rich fines that escaped in stoping.

Open Stoping

Where all the material is removed as broken and the walls of an orebody will stand without support except casual stulls or pillars, the open-stope method is usually followed. This method is used in veins of any degree of dip. In steeply dipping deposits stulls are used for holding drilling platforms. The ore is allowed to drop directly to the bottom of the stope, or it may be directed to chutes by slides held in place by stulls. As the dip decreases, relatively more stripping is required in narrow veins to provide headroom and getting the ore out of the stopes is more expensive; these disadvantages may be offset partly by greater ease in getting around in a stope and less time required to build drilling platforms.

In flat deposits cars or wheelbarrows may be used for removing ore from stopes. On dips over 8° to 10°, however, difficulty is experienced in pushing empty cars up the grade. Broken ore readily runs down slopes of 45° or more unless it is sticky or becomes packed in chutes. On slopes of 20° or 25° to 45°, the ore can be pulled readily or assisted down the slope by means of a shovel. Slopes between 10° and 20° present the most difficult operating problems. The ore either has to be shoveled several times to get it out of a stope, or some mechanical means must be employed. Scrapers are generally used now in base-metal or relatively large-scale gold mines on all slopes down which the ore will not run by gravity. In small-scale gold mines such installations generally are not justified. Moreover, if the ore is high-grade, the probability of losing values may preclude the use of scrapers. Generally, high-grade ore is lowered down flat slopes in a car. The car is pulled up the grade by means of a hoist or a hand windlass which is also used as a brake in lowering.

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[5] Johnson, C. H., and Gardner, E. D., Cut-and-Fill Stoping: Information Circ. 6688, Bureau of Mines, 1933, 58 pp.

Figure 4 shows the general plan commonly used in stoping narrow, flat-dipping gold veins. When stripping or sorting is done the waste is stored in the stope. Where the space is entirely filled the method would be classed as cut-and-fill stoping. Details are varied to suit local conditions. In figure 4 the chutes and manways take up about one third of the stope areas; only two thirds of the space is therefore available for storing waste obtained in stripping the ore.

The ore chutes may be spaced from 20 feet apart, as shown in figure 4, to twice this distance. In orebodies less than 40 or 50 feet long only one ore chute and manway probably would be used. Where the lateral dimension of the deposit is extensive, pillars are left at the drifts. Lean portions of the veins may also be left as pillars. The drift pillars generally are extracted as the last phase of the stoping. In figure 4 stulls are shown only along the chutes and manways; they also may be used wherever they are required for roof support in a stope. In heavy ground the waste would be packed tightly under the back. In such ground, stulls with headboards would be placed at regular intervals as room was made at the face.

Drilling may be done by hand or by air drills. In the latter case stopers are preferred in most mines where the dip is over 20° or 25°. In flatter deposits mounted machines or hand-held jackhammers are generally employed, depending upon the character of material drilled. Figure 5 shows a drilling plan in a face containing 18 inches of ore and 3 feet of stripping. Where practicable the waste is drilled and stripped from over the ore. The waste is then pulled or shoveled into the gob. Planking is placed to prevent waste rolling into the ore chutes or manways. After gobbing the waste the ore is broken and shoveled into the ore chute. The floor of this part of the stope is then cleaned to recover any particles of gold.

It is common practice to sweep the footwall and then wash the walls and timbers with a hose to recover fines in open stopes after the regular stoping is completed. An appreciable amount of gold may be obtained in these clean-ups. The higher recovery thus possible in open stopes makes this method of stoping preferable where other conditions are equal.

The ore chutes usually consist of two rows of stulls lined on the inside by one or two courses of planking. The ore is worked down the chutes by shovels shown in figure 4. If the dip is less, track may be laid in the chute compartment and cars pulled into the stope for loading, or small skips may be used which dump automatically in pockets at the drift.

In nearly horizontal deposits the stopes are laid out to fit the haulage system, and a method analogous to room-and-pillar stoping is used. Usually where stripping is necessary the waste is gobbed to support the back and

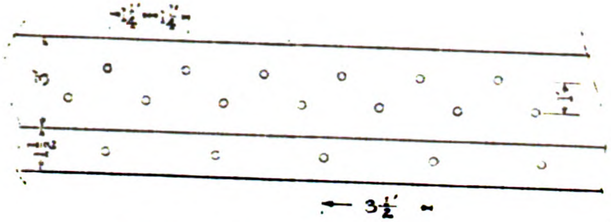


Fig. 5.—Spacing of drill holes in face of stope with 18 inches of ore and 3 feet of stripping.

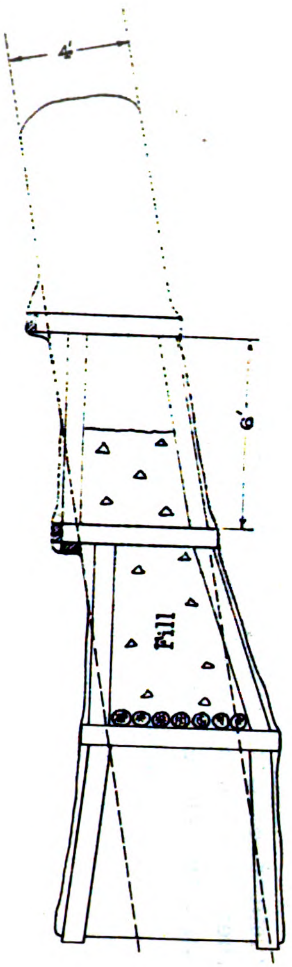


Fig. 6.—End section, square-set stope, with vertical sets.

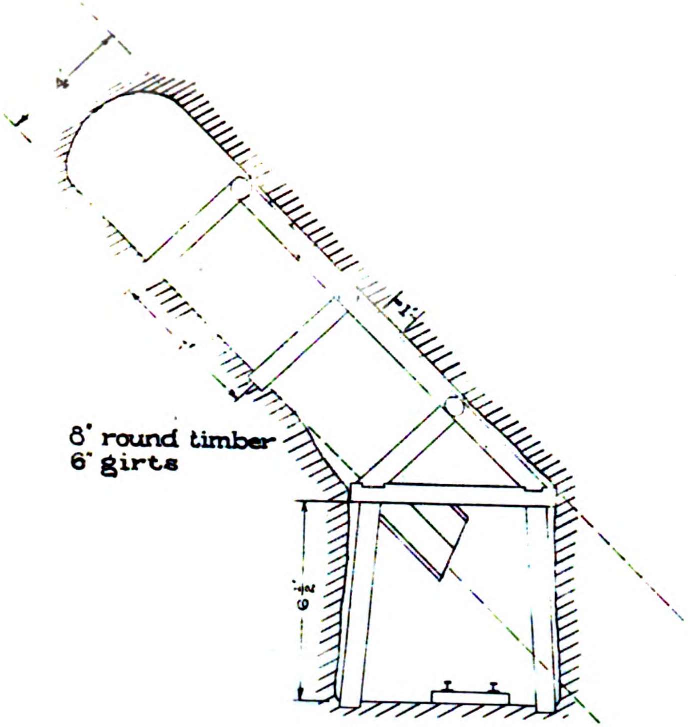


Fig. 7.—End section, square-set stope, with posts normal to walls in narrow vein.

relatively few pillars are required. Where stripping is not necessary the method of stoping (fig. 4) would probably be varied by leaving pillars between stope sections until other stoping operations are completed.

Shrinkage

In shrinkage stoping only enough ore is drawn to allow for the expansion or, as it is called, "swell" of the ore. About one third of the ore is thus drawn as broken; the other two thirds is removed when breaking has been completed. The method is not used in veins dipping less than 45° or less than 3 feet thick. Ore has a tendency to "hang up" in narrow stopes which makes this method of mining dangerous in narrow veins. Moreover, in most small-scale mines expediency requires the ore to be removed as soon as broken to keep a mill running or to provide money to meet the pay roll. Under these conditions open stoping rather than shrinkage stoping would be followed.

Drilling and blasting are done in much the same manner as in cut-and-fill stopes. No sorting or selective mining is, of course, possible in a shrinkage stope. Horses of waste, however, may be left unbroken as pillars. A manway is usually carried up each end of a stope. Chutes are built at intervals of 10 to 25 feet along the bottom of the stope; the ore is drawn into cars.

Shrinkage stopes should be drawn down evenly. The drawing should be done with care; if the stope "holds up" no one should be allowed over the place where this occurs until the obstruction is overcome. Shrinkage stoping has been described in detail by Charles F. Jackson in United States Bureau of Mines Information Circular 6293, entitled "Shrinkage Stoping."

Timbered Stopes

As stated previously, timbering may be necessary to hold stopes open. In nearly vertical veins square-sets or stringer sets may be used with vertical and horizontal members. This method of timbering permits both walls to be lagged readily. In narrow veins dipping 45° or less timbers are usually placed normal to the walls; in nearly flat veins vertical posts are used. In heavy ground the timbers are placed at regular intervals. The hanging wall can be lagged with headboards when stulls are used, but usually when this support is needed sets are preferable. Figure 6 shows a square-set filled stope that dips 80°. The stope may or may not be filled, depending upon whether waste is available or filling is required to support the walls. Where filling is not used and the walls have a tendency to spall or slough, lagging should be used back of the sets.

Figure 7 shows a timbered stope in a vein dipping 45°; it also shows top lagging. If the stopes were filled the lagging probably would not be used. Square-setting has been described in detail in United States Bureau of Mines Information Circular 6691, entitled "Square-Set System of Mining," by E. D. Gardner and W. O. Vanderburg.

TABLE 2. - Stoping data in representative narrow gold veins

Mine	Location	Dip of vein, degrees	Thickness of ore, feet	Thickness of vein, feet	Minimum width of stopes, feet	Method of stoping	Method of drilling	Sorting in stopes	Sorting at surface	Ore produced per man-shift in stoping, tons
Gold Bug.	Rock Creek, Mont.	70	Bunches	3.0	3.0	Open stope	Hand	No	Yes	0.75
Mountain View....	Garnet, Mont.	60	Bunches (0.25 foot to 2 feet wide)	3.0	3.0	do	do	Some	Yes	.50
Guymas...	do	15	1.0	1.0	5.0	Cut-and-fill	do	Yes	No	.25
Drumlummn	Marysvale, Mont.	70	1.5 to 4.5	1.5 to 4.5	4.0	Underhand	do	Yes	No	.75
Paymaster	Townsend, Mont.	90	Bunches	2.5 to 3.0	2.5	Open stope	do	Yes	No	1.50
Hard.Cash	Rodersberg, Mont.	80	0.33	3.5	3.5	Cut-and-fill	do	Yes	No	.08
Southern Cross...	Georgetown, Mont.	75	10.0	10.0	10.0	Square-set-and-fill	do	Yes	No	2.70
Gold Coin	Silver Lake, Mont.	70	4.0 to 7.0	4.0 to 7.0	4.0	do	do	No	No	3.50
Wedge....	Marysvale, Utah	70	0.25	4.0	2.5	Cut-and-fill	do	Yes	Yes	.10
Jewell...	Butte, Mont.	70	0.67	3.0	3.0	Stull set-and-fill	do	Yes	No	.41
Palmer...	Gold Hill, Utah	68	1/ 1.0	1/ 2.2	2.2	Open stope	do	Yes	No	1/ 1.00
Chiefton.	Myrtle Creek, Oreg.	70	1.0	....	....	do	Air	Yes	No	1.20

1/ Where worked by hand.

Stoping Data

Table 2 shows tonnages per man-shift produced in stoping at 13 representative mines where narrow veins are mined.

EQUIPPING SMALL GOLD MINES

The first work on a gold prospect is usually done with hand tools. For a two-man crew digging opencuts and drifting or crosscutting, the following hand tools will be required. The prices listed are quotations of a mining supply house in an Arizona mining district and on a main-line railroad. The prices probably are average for western districts.

Mining hand tools required for two-man crew.

Number required	Unit	Unit price	Total
2	Picks (without handles)	$0.80	$1.60
2	Pick handles	.50	1.00
2	Shovels	1.25	2.50
1	Double jackhammer (8 pounds)	1.35	1.35
1	Hammer handle	.50	.50
2	Single jackhammers (4 pounds)	1.00	2.00
2	Hammer handles	.15	.30
4	Sets of hand steel; 12-, 24-, or 36-inch lengths; 2 1/4 pounds per foot, 27 pounds per set at 16 cents per pound cut in lengths, per set	2.16	8.64
1	Wheelbarrow (steel)	7.50	7.50
1	Blower (complete)	35.00	35.00
1	Anvil (40-pound, cast-iron)	7.25	7.25
1	Sharpening hammer	1.65	1.65
1	Hardy	.50	.50
1	Pick-up tongs	1.50	1.50
1	Saw	1.75	1.75
1	Ax	2.00	2.00
2	Carbide lamps	1.00	2.00
1	12-inch gold pan	.50	.50
1	Mortar and pestle (2-quart)	5.00	5.00
Total		.....	82.54

When the face of an adit reaches more than 100 feet from the surface, a car and track for removing the broken rock will be an economical investment. A 12-cubic foot car will cost about $50; 8-pound rails will cost about $68 per ton and 12-pound rails $67, or $18.10 and $26.80 per 100 feet of track, respectively. For each 100 feet of track 100 linear feet of 4-inch by 6-inch ties would be used. At $55 per 1,000 board-feet, this would be $11. The cost per 100 feet of track for ties and 8-pound rails would therefore be $29.10.

The minimum quantity of mining supplies to be purchased will be:

1	50-pound box of 7/8- by 8-inch, 40-percent strength gelatin dynamite	$ 9.50
1	box no. 6 blasting caps	2.10
1	coil (100 feet) fuse	1.10
100	pounds blacksmith coal	1.50
	Carbide, track spikes, nails, and other incidentals	3.00
	Total initial supplies	$17.20

In sinking shafts by hand a windlass rope and bucket will be required for hoisting the broken rock. A 1 3/4-cubic foot sinking bucket will cost about $9; 1-inch manila rope (3 3/4 feet per pound) costs about 35 cents per pound. Windlasses generally are made on the ground. A drawing of a windlass is shown in figure 8.

As depth is attained at a prospect a gasoline hoist will probably be used. A 3-hp. gasoline hoist (as quoted delivered at Arizona points) will cost about $420 and a 6-hp. hoist $600. A headframe for sinking or for small-scale production is shown in figure 9. For increased production or sinking large shafts to a relatively great depth a larger hoist and a higher headframe would be required. The headframe shown does not permit the use of a cage.

As the scope of operations increases an air compressor and air drills will be desirable. A 120-cubic foot (one-drill) gasoline-driven portable compressor will cost about $1,650 delivered and a 180-cubic foot (two-drill), $2,450 delivered.

When a compressor is installed, a water-supply line, pipe lines, and other equipment are usually found necessary. One-inch pipe costs about $9.67, 2-inch $21.06, and 3-inch $43.55 per 100 feet in small lots from local dealers. If electric power is installed, expensive transformers and switchboards are required.

The price of air drills varies with the model, make, and number bought. The following quotations were given by an Arizona supply house for wet drills for immediate delivery: Jackhammers, $175; light drifters $370, and self-rotated stopers $350.

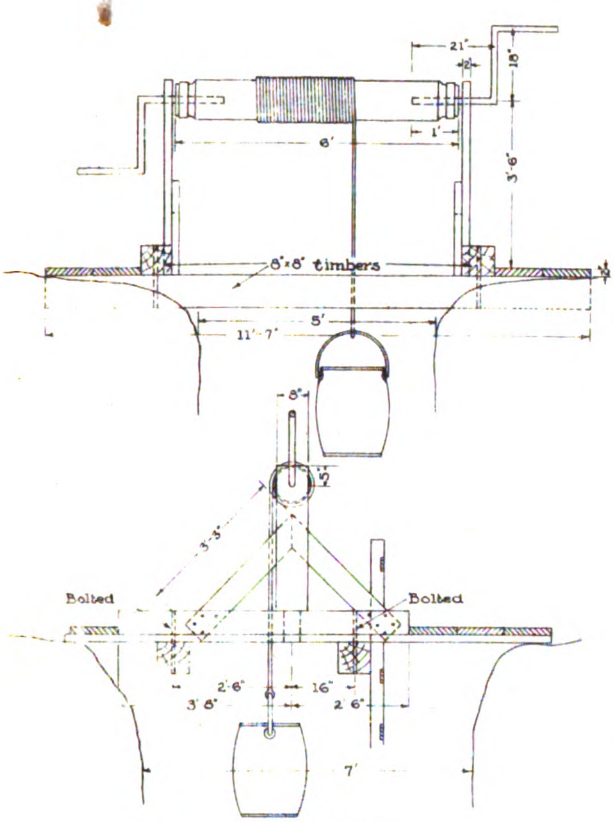


Fig. 8.—Hand windlass.

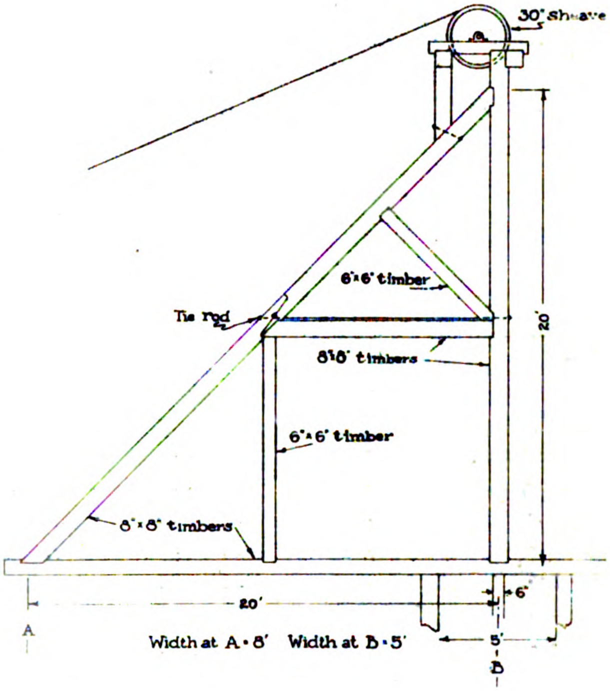
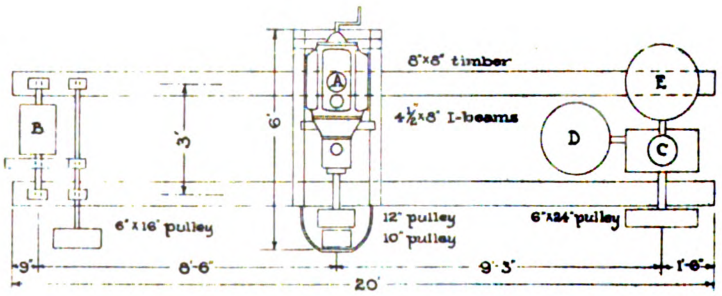


Fig. 9.—Headframe at prospect shaft.



A - V-8 automobile engine
B - 12"x16" geared logging hoist
C - 8"x20" 1 drill compressor
D - 2' x 2½' cooling water barrel
E Air receiver
Assembly mounted on concrete foundation.

Fig 10.—Set-up for driving hoist and compressor from automobile engine.

Drill steel costs about $14 per 100 pounds in small lots. One-inch hollow round steel for stopers or drifters weighs 2 1/2 pounds and 7/8-inch hollow hexagon steel for jackhammers 2 pounds per foot.

In average ground 4 sets of 4 pieces each will be required to drill a round. If two shifts are worked per day and the steel is sharpened on only one shift, 16 sets would be required for one machine. The pieces usually are 1 1/2, 3, 4 1/2 and 6 feet long, respectively, or a total of 15 feet. Sixteen sets of the former would weigh 600 and of the latter 480 pounds. The initial purchase for a one-machine mine is generally 1,000 pounds and if two machines are to be used about 1,500 pounds. A thousand pounds of steel at 14 cents per pound would be $140.

During the past 3 or 4 years small gold operators have installed very few new compressors, hoists, or other mine machinery. Second-hand or homemade equipment has generally been utilized.

An inexpensive set-up for a small shaft mine in Utah is shown in figure 10. The machinery, consisting of a power plant, compressor, and hoist, were set on 8- by 8-inch stringers resting on a concrete foundation. The power plant consisted of a used V-8 automobile engine which cost $100 set up. The engine rested on two 8-inch I beams set across the stringers. The crank end of the engine was held by two 3- by 6-inch H beams welded to the I beams. The other end of the I beams was joined together by a 1/2- by 4-inch steel member which also supported an added bearing for the shaft. A governor on the shaft controlled the speed of the engine through the timing gears. The compressor and hoist were not used at the same time. The engine was fitted with two pulleys on the same shaft. The compressor was driven by a belt from a 12-inch pulley; when the hoist was used the belt was moved to a 10-inch pulley. The compressor had a one-drill capacity and cost $350 new 7 years ago. The air receiver consisted of an oil drum. The cooling water discharged into a barrel and thence went through an old boiler for cooling.

The hoist was a geared logging machine and had a 12- by 16-inch drum which received a 3/8-inch cable; it cost $214 new several years ago.

About 70 buckets were hoisted from the 160-foot level in a day, using 3 gallons of gasoline. Four gallons of gasoline was required to run the compressor 1 1/2 hours, the length of time required to drill a drift round. About 10 buckets of muck were broken to a round.

Similar installations powered by old automobile engines were noted throughout the western districts. At a small mine near Butte, Mont., the hoist consisted of an old hand winch rigged up to be run by an old automobile engine. A band on a pulley attached to a shaft served as a brake. The cost of this installation was less than $100. The headframe consisted of two 8- by 8-inch by 16-foot vertical timbers with crosspieces to hold the sheave; these were braced by two 8- by 8-inch inclined members, and the whole assembly was guyed with old hoisting cable. Development work was being done and ore mined on the 100-foot level. The ore and waste were hoisted in an 8-cubic foot bucket. One surface man ran the hoist, tended the compressor, and took away the car into which the muck was dumped. He also sharped the steel used underground.

TABLE 3. - Cost of equipping representative small gold mines

Mine	Location	Daily tonnage	Depth of shaft	Main items of equipment	Total cost
Eureka Standard	Dividend, Utah	100	1,400	85-foot steel headframe, hoist, 20-drill compressor, 12 pumps, steel ore bin, storage battery locomotive.	$299,500
Yankee	American Fork, Utah	40	None	Nine-drill compressor,1/ $5,000; 4.3-mile tramway, $100,000	....
Palmer	Gold Hill, Utah	12	300	Two-drill compressor, $1,500; 3.0-hp. gasoline hoist, $900, 6-hp. gasoline hoist, $200; one-drill compressor,1/ $300; blacksmith shop, including drill sharpener, $800; 110-ton ore bin, $450; assay office, $1,500; camp buildings, $2,675.	8,325
Sheep Rock	Beaver, Utah	Drifting	300	Hoist, $214; compressor, $350; engine,1/ $100; foundations, car, and incidentals, $150.	814
Gold Queen	do	Sinking	60	Blower,1/ $15; anvil, $14; two tents,1/ $30; 100-foot cable,1/ $2; 100 pounds steel, $14; hand tools,1/ $8.	83
Record	Unity, Oreg.	Drifting	None	340-cubic foot compressor with gasoline engine, set-up, $3,500.	3,500
Gnome	Orogrande, Idaho	25	None	Blacksmith shop, cars, tracks, hand and miscellaneous tools for 30 men, etc.	2,500
McKinley	Lucile, Idaho	Drifting	None	210-cubic foot compressor, gasoline-engine drill sharpener, $500; Leyner drill, $330; drill steel, $330.	....
Cane Springs	Gold Hill, Utah	15	200	220-cubic foot portable compressor1/ and two new drills, $2,200; 20-hp. hoist,1/ 500-foot cable, 16-cubic foot bucket, $100.	2,300
Wedge	Marysvale, Utah	1	None	Two 1/2-ton cars, hand tools for two men, Hartz hand jig.	200

1/ Second-hand.

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The cost of equipping 10 representative small gold mines is shown in table 3.

MILLING

The ore from most gold mines that produce regularly over 25 tons daily is milled rather than shipped to a smelter. Milling costs are usually less than smelter charges, and further savings in trucking and freighting are made. Moreover, smelting companies usually pay for only about 90 percent of the gold in the ore; when bullion is shipped to the mint the full value is received by the shipper.

Three general types of gold mills are commonly used: (1) Amalgamation-gravity concentration, (2) flotation, and (3) cyanidation. In the first type the crushed and ground ore is passed over amalgamation plates and then over concentrating tables. The products obtained are an amalgam from which the gold is obtained as bullion at the mine and a table concentrate which is shipped to a smelter. A concentrate which requires smelting is obtained by flotation and a gold precipitate by cyanidation. Combinations and modifications of these general types of treatment are used about as often as a simple process.

Choice of Methods

The major considerations for the selection of the type of mill at any given property are: (1) Nature of the ore (including the kind of ore and gangue minerals and the amenability of the ore to the several methods of treatment); (2) amount of ore developed in the mine; (3) relative cost of treatment by the different methods; (4) cost of marketing the products of treatment; (5) initial cost of milling plants of the different types; (6) capital available.

Free-milling gold ore is the only class that is well defined; a large proportion of the gold in this ore is released by relatively coarse grinding in such a condition as to amalgamate readily. From such ores it may be possible to recover 50 to 70 percent of the gold by amalgamation after crushing through perhaps 40-mesh screens. By tabling the ore after it leaves the plates a concentrate may be obtained and the total recovery increased, but rarely above 75 to 85 percent. Free-milling ore is characteristic of shallow, oxidized zones. Often when mining extends below the water level the ore is found to be more difficult to treat by amalgamation. Some gold ores contain minerals that affect amalgamation adversely and may prevent their treatment by this method. Minerals containing arsenic, antimony, lead, bismuth, talc, clay, or graphite may cause a dull surface film to form on the mercury and thus retard amalgamation of the gold. When this occurs recovery drops and mercury loss may become excessive. The gold in some ores that might otherwise be free milling is "rusty" and will not amalgamate. In general, however, if an ore is not free milling, it is because most of the gold is in such fine particles that it does not amalgamate readily, even when set free by fine grinding.

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Almost all gold ores can be treated by either flotation or leaching in cyanide solution, or both, with recoveries generally higher than those by amalgamation. Neither process recovers coarse gold readily; therefore, if the ore contains appreciable amounts of gold that will not pass, say, a 30-mesh screen it is customary to supplement flotation or cyanidation by tables, plates, or other devices to recover this gold. Fine gold particles, however, float readily and dissolve readily in cyanide treatment, so that the choice between the two methods usually is based on relative recoveries, operating costs, and initial costs, rather than on the practicability of either method. Recoveries by cyanidation generally are higher than those by flotation, ranging from 90 to 98 as compared with 80 to 90 percent.

Ores that contain appreciable quantities of cyanicides—that is, minerals or foreign materials that dissolve in a cyanide solution and hence consume cyanide—cannot be treated by the leaching method. Oxidized copper minerals, arsenides, and various basic salts are such elements. Graphite or organic materials precipitate gold from solution and thus prevent extraction. Ores that contain much clay or that yield excessive amounts of slime on grinding may offer difficulties in cyaniding by making the pulp hard to settle or filter.

Rusty, coated gold is not readily floated. Flotation applied to ores containing large amounts of sulphide may give a low ratio of concentration and a corresponding low-grade concentrate. Only rarely can differential flotation be applied to correct this condition, as usually all the sulphides are gold-bearing and must be saved.

The amount of ore developed is the major factor influencing the scale of operations and thus bears on the choice of methods. The capital expenditure on a cyanide plant may prohibit its application to very small tonnages except under special conditions. Furthermore, considerable expense is involved in ore testing, sampling, and assaying in the operation of cyanide plants (also flotation plants) which greatly increases the treatment cost in small installations. Amalgamation and tabling, on the other hand, are relatively simple processes less affected by "rawhide" methods of control. If the ore reserves are large, mining usually can be controlled to produce mill feed of uniform grade and character, which is desirable for flotation and cyanide work but less important for amalgamation and gravity concentration.

Treatment by amalgamation and tabling in general is the cheapest of the three principal methods. The relative operating costs of flotation and cyanidation cannot be generalized, although under given conditions one may be far cheaper than the other. Grinding is the most expensive phase of most milling operations; the amount of grinding necessary to effect the greatest profit has an important bearing on the operating cost of each method of treatment. As relatively coarse feed is satisfactory for amalgamation, grinding is seldom carried below 40- or 60-mesh. Generally, finer grinding is required for flotation than for cyanidation, but many exceptions to this are found. At Oatman, Ariz., for instance, grinding to 85 percent minus 200-mesh precedes cyanidation. Cyanidation, being the most complex and

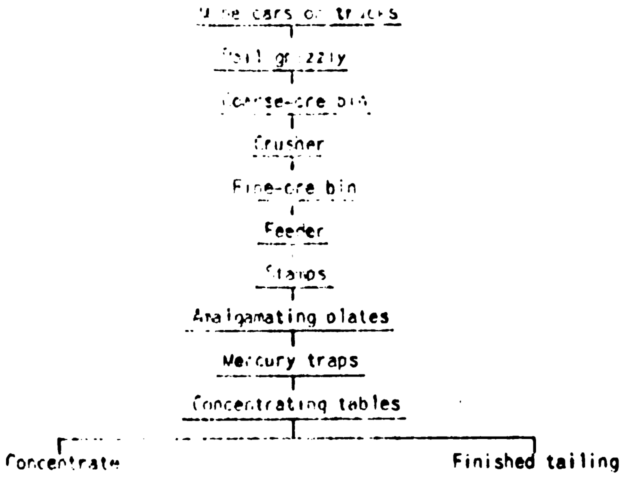

Fig. 11.- Flow sheet of typical amalgamating-tabling mill.

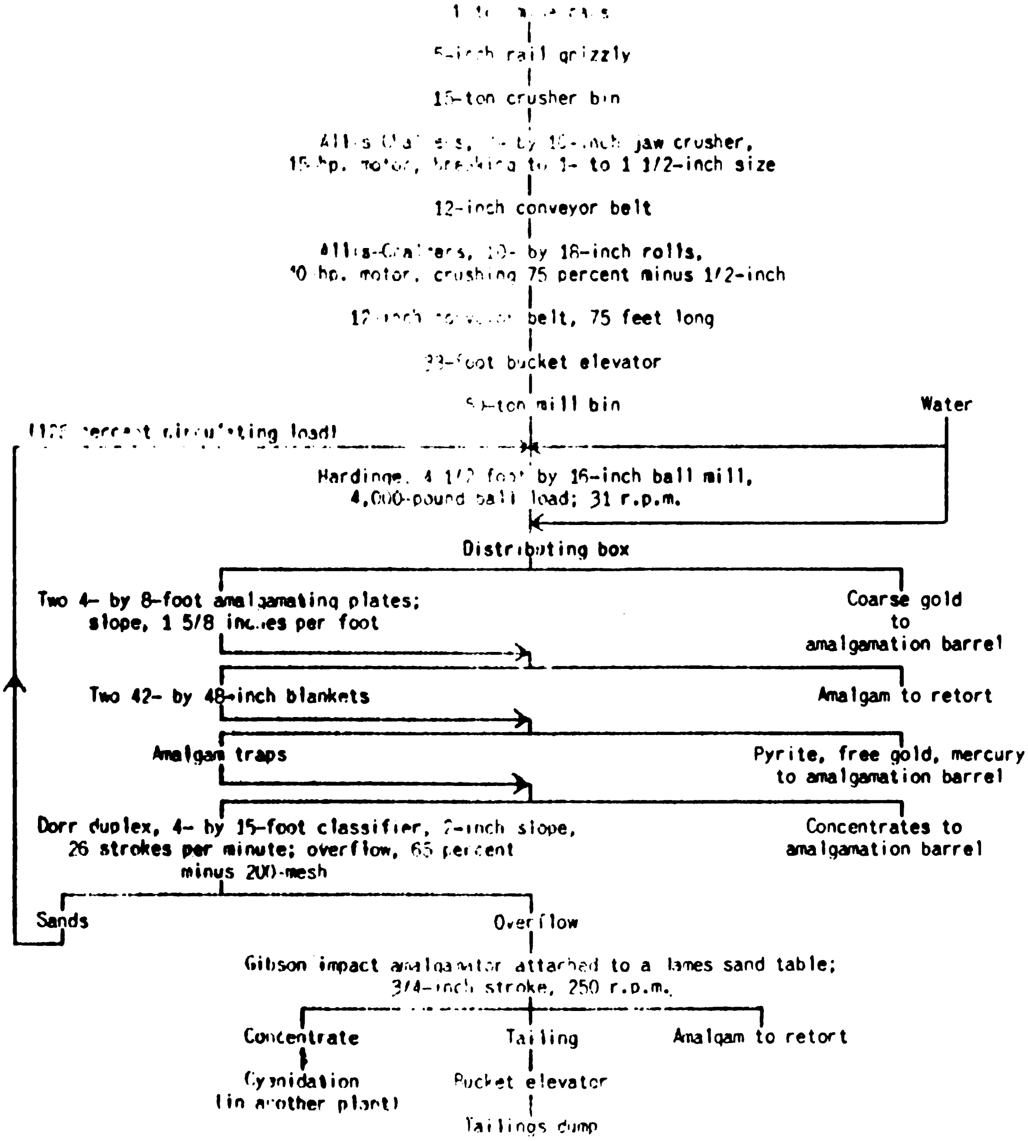
1-ton mine cars
5-inch rail grizzly
15-ton crusher bin
Allis-Chalmers, 7- by 10-inch jaw crusher,
15-hp. motor, breaking to 1- to 1 1/2-inch size
12-inch conveyor belt
Allis-Chalmers, 10- by 18-inch rolls,
10-hp. motor, crushing 75 percent minus 1/2-inch
12-inch conveyor belt, 75 feet long
33-foot bucket elevator
50-ton mill bin
Water
(125 percent circulating load)
Hardinge, 4 1/2-foot by 16-inch ball mill,
4,000-pound ball load; 31 r.p.m.
Distributing box
Two 4- by 8-foot amalgamating plates;
slope, 1 5/8 inches per foot
Coarse gold
to
amalgamation barrel
Two 42- by 48-inch blankets
Amalgam to retort
Amalgam traps
Pyrite, free gold, mercury
to amalgamation barrel
Dorr duplex, 4- by 15-foot classifier, 2-inch slope,
26 strokes per minute; overflow, 65 percent
minus 200-mesh
Concentrates to
amalgamation barrel
Sands
Overflow
Gibson impact amalgamator attached to a James sand table;
3/4-inch stroke, 250 r.p.m.
Concentrate
Tailing
Amalgam to retort
Cyanidation
(in another plant)
Bucket elevator
Tailings dump

Fig. 12.- Flow sheet of Porcupine United Gold Mines 25-ton amalgamation plant.

technically difficult process, requires the most skillful operators. Moreover, cyanidation in small plants requires more labor--that is, more man-hours per ton treated--than flotation. Flotation, in turn, requires more and better-trained operators than amalgamation.

Amalgamation and cyanidation produce bullion that usually is marketable at a nominal cost for the full value of its gold and silver content. Flotation and tabling, on the other hand, yield concentrates that must be marketed; this expense may be 10 to 15 percent of the entire milling cost, in addition to appreciable loss of gold in smelting. This factor is of greatest importance in remote districts where transportation to smelters is costly. This disadvantage of flotation can be only partly remedied by producing a rich concentrate, as the saving in the tonnage to be transported is offset partly by increased freight rates and smelting charges. Other methods of approach to the problem are to amalgamate or cyanide the concentrates.

The initial cost of well-designed, small gold mills has ranged from a few hundred dollars to $2,000 per ton of daily capacity, depending on where they were built, whether new or used machinery was installed, the size of grinding unit needed to give the required capacity, and the type of plant. The amalgamation and tabling plant is the cheapest, followed in order by flotation and cyanidation plants.

The effect of limited capital is seen repeatedly in the use of the type of plant having the lowest first cost although another might yield a greater ultimate profit. The same effect usually extends to the design, construction, and choice of machinery in the mill.

Amalgamation and Gravity Concentration

Figure 11 shows the generalized flow sheet of a typical small amalgamation-tabling plant. Occasional boulders that would make drawing from the coarse-ore bin difficult or that would be too large to enter the crusher are broken by hand on the grizzly.

The coarse-ore bin is generally of timber and of conventional sloping bottom with a chute in front. The bottom may be covered with sheet steel or with a renewable board or plank lining. Capacities range from a few to 20 tons. Crushing usually is confined to day shift; the coarse-ore bin, therefore, should be large enough to hold all ore brought from the mine during the night shift.

Coarse crushers are generally of the Blake or Dodge type and commonly are fed by gravity through a wooden chute with sliding plank gates. In very small plants the ore may be dumped on a platform and shoveled by hand into the crusher. The crushers usually range in size from 6 by 9 to 10 by 20 inches and are set to break to minus 1 1/2 inches to 3/4 inch. Two-stage crushing is seldom necessary or economical; gyratory crushers are unsuitable for the type of plant under discussion. In a plant where individual electric drives are not used, power for the crusher is often provided by a separate engine.

In small mills where the building site permits, the crusher may be on top of the fine-ore bin. More often elevators or belt conveyors carry the ore from the crusher to the fine-ore bin. In a few plants where custom ores are handled, weighing and sampling equipment is installed between the crusher and the bin. Otherwise, tonnages are usually calculated from the number of carloads or truckloads and samples taken manually just ahead of the grinding unit.

The fine-ore bin is usually of timber construction and likewise of the sloping-bottom, front-discharge type. Its capacity should be 1 or 2 days' mill run, as this will make the mine and mill fairly independent; moreover, a large bin permits better mixing of the ore and thus provides a more uniform mill feed than a bin of small capacity.

Various types of feeders are in use. The Challenge disk-type feeder is a favorite with sloping-bottom bins and stamps. However, small corrugated drum feeders, reciprocating plate feeders, shaking chutes, plunger feeders, and belt- or apron-type feeders are in common use. If properly installed all of these work satisfactorily.

Stamps are shown in the flow sheet only because of the conventional association of stamps and amalgamation. Ball and rod mills work satisfactorily in plants of this type. Important features of the stamp mill are its use as an amalgamator and the relative ease with which it is cleaned out. On the other hand, a ball mill acts as a trap for coarse gold, and the plant must be shut down for several hours when it is cleaned out. Quicksilver is sometimes used in a ball mill, but the danger of loss by flouring offsets the increased chance for amalgamation.

Five-, ten-, or twenty-stamp mills are usual, and the stamps range from 800 to 1,100 pounds in weight. Capacity ranges from 3 to 6 tons per stamp per 24 hours, grinding to the usual size for plate amalgamation. The screens used are both the woven-wire and punched-plate type, the latter often having slotted openings. The proper size of screen can be determined only by test; 20 to 40-mesh screen cloth is in common use. A product as coarse as 10-mesh causes serious scouring of plates. Crushing finer than 20-mesh may prove uneconomical, as the higher cost and lowered capacity may nullify any increase in recovery. The power needed to operate stamps of the sizes commonly used in small mills ranges from 2.0 to 2.5 hp. per stamp.

Aside from the use of quicksilver in the stamp batteries, amalgamation is commonly done on plates in the type of mill under discussion. Other amalgamating devices, such as table amalgamators, are in use, but as no sound basis for comparison with plates was found their value must be left an open question. One plate or series of plates is provided for each 5-stamp battery. The plate area varies widely, but 50 square feet per battery is an approximate average. If the gold is very fine, more plate area is desirable. The plates are best cut into sections with 1- or 2-inch drops between and a similar drop provided where the pulp comes on the plate. The slope of plates ranges from 1 1/2 to 2 inches per foot. The care of plates influences recovery markedly; if the amalgam surface is not kept bright and soft, gold is lost. Plates are cleaned with cyanide or caustic soda and dressed with quicksilver at 4- to 6-hour periods; the clean-up period varies more widely but is often 24 hours.

Mercury traps and riffled sluices are installed below the plates principally to catch rusty gold, floured mercury, and loose particles of amalgam passing over the plates. When the plates do not function correctly a considerable saving of gold may be made in the traps.

The main purpose of tabling after amalgamation is to save the sulphide content of the ore or heavy oxidized minerals that contain gold or can be shipped profitably for their own value. The concentrate is usually caught in settling boxes, shoveled out periodically, dried, and sacked. The operating cost of a table is very low, and its water consumption is slight when treating the dilute pulp from amalgamating plates. Because of the relatively low recovery by amalgamation, the tailings from a mill of this type should be stored, if feasible, rather than run to waste, with a view to possible future treatment.

A 5-stamp mill of the type described was built in California in 1931-32 for $10,000. Electric power was available, so the cost excluded a power plant. The installed motors totalled 20 hp. The stamps, weighing 1,000 pounds and dropping about 6 inches and 110 times a minute, had a capacity of 20 tons in 24 hours through a 40-mesh screen.

Three amalgamators and a crusherman on day shift usually constitute the crew for a small mill of this type. If tailings disposal or power-plant operation is included, a crew of 6 to 8 men may be necessary. The labor cost may range from $0.50 to $2.00 per ton and total operating cost from $1.00 to $3.50 per ton.

Modification of the simple flow sheet described that yet retain plate amalgamation as their chief method of recovery are numerous. One such may be described briefly. At the Porcupine United Gold Mines, Ltd., Timmins, Ontario, a 25-ton amalgamation plant was put in operation in March 1929.5/ The flow sheet is shown in figure 12, in which the points of difference from figure 11 are apparent. The table feed was 65 percent minus 200-mesh, but it is to be noted that the amalgamation plate feed was the unclassified ball-mill discharge. Milling had to be controlled to prevent the discharge of material so coarse that it would scour the plates. As this discharge contained about 35 percent solids, water was added to it ahead of the plates. Forty percent of the gold recovered was taken from the plates, which were dressed every 3 hours and cleaned daily. The distributing box acted as a trap for coarse gold and was cleaned out periodically.

5/ Vary, R. A., Amalgamation Practice at the Porcupine United Gold Mines (Ltd.), Timmins, Ontario: Information Circular 6433, Bureau of Mines, 1931, 5 pp.

The blankets, composed of what is known as "silence cloth", were washed out in a tub every time the plates were dressed; the concentrates from here, together with material periodically cleaned out of launder, sumps, and distributing box, were ground in a cast-iron amalgamation barrel. The treatment in this barrel consisted of 10 hours grinding with steel balls, followed by addition of mercury and lime and several hours more grinding. Then the barrel was washed out, the amalgam and mercury were collected and washed, and the residue was returned to the mill. About 35 percent of the recovery was made in this fashion.

The amalgamator on the James table caught fine gold and float mercury; it was cleaned weekly. The table concentrates, averaging $40 per ton, were dewatered and sent for cyanidation to a neighboring mill.

The mill crew included a crusherman and a foreman operator on day shift and one amalgamator on each of the other two shifts. The ball mill and other machinery in the mill proper were driven from a line shaft by a 50-hp. motor, except the classifier which was driven by a 5-hp. individual motor. Power was purchased at a cost of about 1 cent per kilowatt-hour.

Mill heads averaged $11.00, amalgamation tailings $2.80, and table tailings $1.80 per ton. Segregated costs per ton are shown in the following tables. Ball and liner consumption was about 2 pounds per ton; 0.05 ounce of mercury was lost per ton; power consumption was 40 kw.-hr. per ton.

Operation costs per ton of ore treated

	Labor	Power	Supplies	Total
Crushing	$0.214	$0.135	$0.035	$0.384
Grinding	.429	.180	.162	.771
Classifying, screening, conveying, and refining	.429	.090	.107	.626
Miscellaneous	......	......	.070	.070
Total	1.072	0.405	0.374	1.851

Flotation

The flotation process has come into extensive use for treating gold ores during the last few years. It is usually supplemented, however, by one of the other methods of concentration. One reason for this is that many ores contain gold particles too large to be floated readily. Another reason is that with a mill feed of the varying character usual in small operations some safeguard or indicator, such as a scavenger table, is needed to warn against

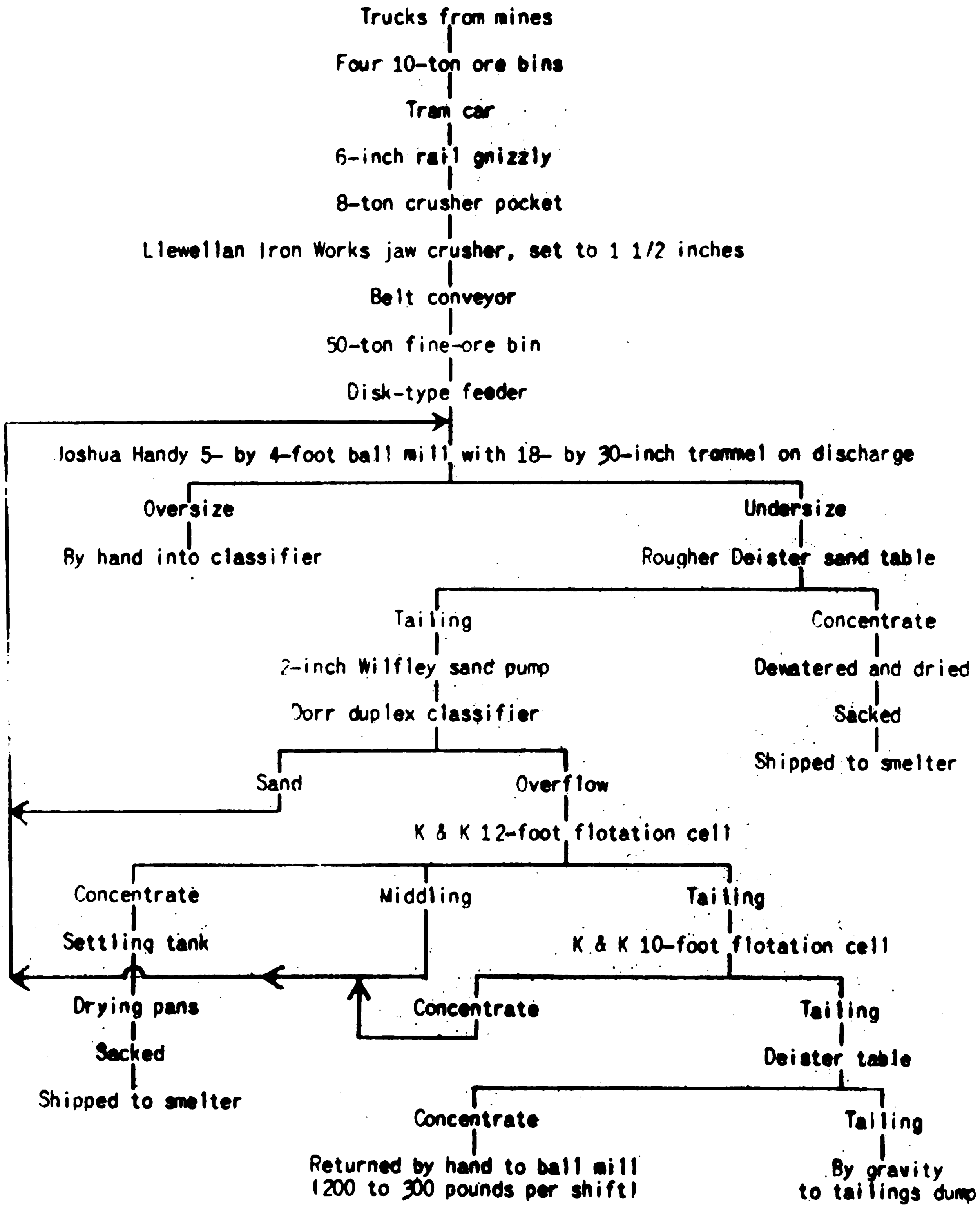

Fig. 13.- Flow sheet of a 35-ton gold flotation plant.

serious losses. Grinding for flotation is usually done in closed circuit with a classifier, and it is often found advantageous to insert a table, plates, or blankets in the circuit to remove free gold, especially the coarser particles that otherwise would settle in the mill or classifier. Incidentally, this practice may permit the marketing of part of the gold in bullion form rather than as a concentrate.

The principles of flotation of gold ores are less well understood than those of amalgamation, cyanidation, or gravity concentration; some of its practices are still in the experimental stage. In any particular case, the reagents, degree of grinding, and type of cell to be used should be determined by laboratory test before designing the mill. Many types and makes of cells are in use, and the results appear to favor none. Grinding is carried farther than for amalgamation or for percolation leaching; 100 percent of the ore, as a rule, is ground to pass 60-mesh and 65 to 85 percent to minus 200-mesh.

The flow sheet of a 35-ton gold flotation plant is shown in figure 13. As ore from several sources was treated, separate small ore bins were provided as well as a patio above the mill where ore could be dumped. A 1-ton mine car was used to tram the ore from the ore bins to the crusher pocket; platform scales were installed so that each carload could be weighed. The jaw crusher was driven by a vertical-type hothead (semi-Deisel) oil engine; the crusher was about 100 feet from the mill ore bin. Samples were taken manually where the crushed-ore conveyor discharged into the fine-ore bin.

The trommel on the ball-mill discharge originally was made of 1/4-inch mesh screen cloth, but a finer size was used when it was found that coarse particles were diluting the table concentrate. The rougher table concentrates were very rich.

The first flotation cell produced a finished concentrate from the first 6 feet of its overflow and a middling from the other 6 feet. The middling joined the froth from the scavenger cell and was pumped back to the classifier.

It was contemplated that the concentrates could be cyanided profitably in a small auxiliary plant of the percolation-leaching, zinc-box type, inasmuch as transportation to smelter and smelter charges were costly and losses were high.

Water was scarce, and the trouble and expense of reclaiming water from small dams on the slope below the mill were at first serious.

The ore was quartz with free gold, galena, pyrite, and arsenopyrite. That from different deposits varied widely in hardness and flotation characteristics. The average mill heads carried about 0.6 ounce of gold and 3 ounces of silver per ton. The combined flotation and table concentrate carried 10 to 14 ounces of gold per ton, 15 percent of lead, and 40 percent of insoluble.

The regular mill crew consisted of 10 men, including a foreman, trammer, crusherman, and sampler on day shift and a flotation operator and tailings-disposal man on each shift. The tailings men also dried and sacked the concentrates.

Except for the crusher, power for the mill was supplied by a 120-hp., 4-cylinder Deisel engine. A small generator for supplying lights at the mill was belted to the Deisel. Two or three small gasoline engines were used on the water supply.

The plant was built in 1933-34 and cost $17,500; nearly all of the machinery was second-hand.

Cyanidation

Cyanide plants are of two types--percolation and agitation. Where the ore is porous and the gold relatively fine, a satisfactory saving of the gold may be made by simple percolation in tanks. This permits a considerable saving in grinding costs. Where the ore is dense, however, fine grinding is necessary in order that the cyanide solutions may come in contact with the gold.

Figure 14 shows the flow sheet of the Big Jim (Telluride) mill at Oatman, Ariz., which is representative of an agitation type of small plant. The rated capacity was 50 tons per 24 hours. The ore treated consisted of a dense quartz-calcite gangue containing free gold. It contained no sulphide but had about 1 ounce of silver to 2 ounces of gold. The gold was so finely divided that unusually fine grinding was necessary.

The flow sheet comprised coarse crushing, 1 stage of fine grinding, and 1 stage of thickening from which the pregnant solution was derived, followed by 3 stages of agitation and 5 stages of thickening from which a finished tailing was obtained. The pregnant solution was clarified and then treated by the Merrill-Crowe process.

The crusher was driven by a 50-hp. motor and was operated 4 to 6 hours per day. The ball mill was driven by a 75-hp. motor at 18 r.p.m. The classifier discharge was 80 to 82 percent minus 200-mesh. Ball consumption ranged from 2.5 to 4.5 pounds of chrome-steel balls per ton of ore. Grinding was done in cyanide solution. About 50 percent of the gold in the ore was dissolved in the mill. The cyanide solution was drawn from the main mill storage tank at the rate of 330 tons per 24 hours. The classifier overflow carrying 15 percent solids flowed to no. 1 thickener which was 24 feet in diameter and 8 feet deep. The overflow passed by gravity to the precipitation section of the plant. The thickened discharge which contained 40 percent solids was raised by a diaphram pump to the first of three agitating tanks. Tank no. 1 was 19 feet in diameter and 14 feet deep; nos. 2 and 3 were 13 feet in diameter and 14 feet deep. Air for the agitator and lifts was supplied at a pressure of 30 pounds per square inch by a 7 1/2- by 6-inch compressor driven by a 10-hp. motor. About 95 percent of the gold had been taken in solution by the time the pulp left no. 3 agitator. From there it went to no. 2 thickener and thence to nos. 3, 4, 5, and 6. The thickeners were 15 feet in diameter and 10 feet deep. The discharge from no. 6 which was the tailing contained 50 percent solids. Fifty tons of flush water was added to no. 6 thickener daily; 180 tons of barren solution from the precipitation section was added to the no. 4 thickener. This solution contained 2.2 pounds of sodium cyanide, 2 pounds of lime, and a trace of gold to the ton. The overflow from no. 2 thickener, 330 tons per day, was pumped to the mill storage tank. The pregnant solution from no. 1 thickener ran to a Butters clarifying filter having a tank 11 feet in diameter and 5 1/2 feet deep; it was then pumped to the clear gold-solution tank 11 feet in diameter and 6 feet deep. From there it passed through the Merrill-Crowe, vacuum-treatment, zinc-dust precipitation process. Zinc dust and lead acetate were added automatically to an emulsifier ahead of a triplex circular pump.

The precipitate press was cleaned twice monthly; the precipitate was placed in a graphite crucible and melted in an oil-fired furnace. For each 100 pounds of precipitate, 20 pounds of borax, 10 pounds of soda, and 6 pounds of silica were used for fluxing.

The total installed electric power in the mill was 200 hp. Nine men were on the mill pay roll, including a superintendent. Cyanide consumption was 0.75 to 1.00 pound per ton of ore and lime consumption 4 pounds per ton. Zinc dust was added at the rate of 8 pounds per 24 hours or 0.7 ounce per ton of solution; lead acetate was added at the rate of 1 ounce per ton of ore.

The recovery was 97 to 98 percent. The milling cost was $2.44 per ton, including a rental charge of $1,000 per month for the mill. The plant is reported to have cost $108,000 in 1923 when it was built.

To reduce the loss of soluble gold in the tailing the recent trend in cyanide practice is to substitute filters for some of the thickeners. The pulp may be filtered once and discharged, or the cake from the first filter may be pulped and filtered a second time.

Where a satisfactory saving can be made by fairly coarse grinding, the sands may be leached by percolating the cyanide solution through the ore in tanks; the ore is crushed by stamps or ground in a ball mill to minus 8- or 10-mesh. The sands are separated from the slimes by classification; the slimes are treated by flotation or leached by agitation. At a few plants treating a porous product the ore is crushed and then reduced to about 1/2-inch size by rolls. Rolls are used to keep sliming to a minimum. The ore is then fed to a series of tanks with filter bottoms. Usually one tank in a series is filled with ore, the tailing discharged from the second, the ore in the third leached with strong solution, that in the fourth leached with weak solution, and the tailing in the fifth washed. The gold is precipitated from the solution and the precipitate handled by the standard method.

The percolation method when applicable has decided advantages in that the cost of the plant is relatively low and the cost per ton of treating the ore is very much lower than that with agitation leaching. The elimination of fine grinding greatly reduces the demand for power. The extraction by percolating leaching, however, seldom exceeds 75 percent, as compared to about 95 percent by agitation.

COSTS

For an ultimate profit to be made in working a mine, returns from the ore must cover the mining and treatment costs and repay all capital expenditures. At small gold mines with limited reserves it is difficult to estimate the charges per ton for amortization and depletion. This is true especially where a property is in the development stage and probably more ore will be opened up. Moreover, any ore that can be produced at less than the direct cost is usually taken out.

Direct costs can be given closely where records are kept. The principal factor governing the stoping cost at the type of mines discussed in this paper is the size of the orebodies.

The amount and cost of development required per ton of ore stoped depends upon the size and distribution of the orebodies. The geological skill exercised in the search for orebodies will also have an important bearing on the amount of prepaid development which ultimately will have to be charged against each ton of ore produced. In many cases, however, where new companies have been formed to take over previously worked mines advantage may be taken of old work that already has been liquidated and for which no charges need be made against future production. Moreover, lessees who have no responsibility for work previously done may work sections of a mine.

Milling charges will depend upon the size of the operation and the type of milling plant. Trucking and freight costs depend mainly upon distances. The costs of supervision and overhead depend upon the form of organization and the quality of the management. Where a small group of men are working on their own account there may be no expense for supervision or overhead. Royalty and smelter rates are matters of negotiations.

Labor is the principal item of expense. Where the work is done by hand, wages may account for as high as 80 percent of the total direct cost.

Segregated costs at 10 representative small gold mines are shown in table 4.

It will be noted that in this group of mines total costs range from $3.74 to $37.10 per ton. The conditions at the mine where the minimum costs are shown were unusually favorable, and a relatively large tonnage (90 tons) was mined daily. Mining costs may greatly exceed the maximum shown in the table where rich ore is extracted from very narrow seams. At a one-man mine in Montana the cost of producing a ton of ore was $1,800 in 1933; the ore contained $2,000 in gold.

TABLE 4. - Costs at 10 representative small gold mines[1]

Mine	Location	Tons per day	Thickness of ore, feet	Type of mill	Mining			
					Stoping	Development	Miscellaneous	Capital charge for equipment and prepaid development
Little Fanny.	Catron County, N. Mex.	90	3	Flotation	[2]$1.81	......	....	....
Big Jim	Oatman, Ariz.	50	5	Cyanide	1.93	$0.71	....	....
Herron and Laster Lease	Superior, Ariz.	50	4	None	[2] 5.31	......	....	$0.66
Gnome	Orogrande, Idaho	20	......	Cyanide	[2] 9.00	......	....	....
Gold Coin ...	Silver Lake, Mont.	7	5 1/2	Amalgamation and tables	2.40	1.09	$0.50	....
Jewell	Butte, Mont.	1 1/4	1	None	10.08	5.04	1.50	2.40
Mountain View	Garnet, Mont.	1	1	None	8.00	8.00	.50	....
Gold King ...	Pinos Altos, N. Mex.	1	1/2	None	[9]15.00	8.50	....	[10]3.00
Gold Bug	Rock Creek, Mont.	3/4	2	Amalgamation	5.90	7.00	....	4.00
Guymas	Garnet, Mont.	1/2	1	None	18.60	([13])	....	....

(See page 30 for footnotes)

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TABLE 4. -Costs at 10 representative small gold mines1/--Continued

Mine	Location	Tons per day	Thickness of ore, feet	Type of mill	Trucking		Milling	Supervision and general expense	Smelting per ton of ore	Freight per ton of ore	Royalties	Total
					Per ton	Distance, miles						
Little Fanny.	Catron County, N. Mex.	90	3	Flotation	0	......	$1.20	$0.73	......	......	......	3/$3.74
Big Jim	Oatman, Ariz..	50	5	Cyanide	$0.30	1	4/2.44	.....	0	0	......	5/ 5.38
Herron and Laster Lease	Superior, Ariz.	50	4	None	.75	1	.00	.50	6/$5.00	$0.16	6/$3.70	15.48
Gnome	Orogrande, Idaho	20	......	Cyanide	0	......	3.00	6/1.50	0	0	......	7/13.50
Gold Coin ...	Silver Lake, Mont.	7	5 1/2	Amalgamation and tables	.30	1/2	1.30	.35	8/ .13	8/ .05	......	7/6.12
Jewell	Butte, Mont.	1 1/4	1	None	1.00	1	0	0	5.00	.83	1.80	27.70
Mountain View	Garnet, Mont.	1	1	None	11.00	110	0	0	5.00	0	......	21.50
Gold King ,,,	Pinos Altos, N. Mex.	1	1/2	None	1.00	11	0	.....	3.50	1.21	......	7/31.10
Gold Bug	Rock Creek, Mont.	3/4	2	Amalgamation	11/2.25	3/4	12/3.70	1.00	0	0	2.25	26.10
Guymas	Garnet, Mont.	1/2	1	None	3.00	12	.00	8.50	5.00	2.00	......	7/37.10

1/ Miners' wages $4.00 or $4.25 per day, except where otherwise indicated.
2/ Includes development.
3/ Excludes capital, royalty, and smelting of concentrates; wages about $2.50 per day.
4/ Includes rental of mill.
5/ Does not include supervision or royalty.
6/ Estimated.
7/ Excludes capital expense and royalty.
8/ Concentrate shipped, ratio of concentration 40 to 1.
9/ Wages at $3.00 per day.
10/ Rental compressor and hoist.
11/ Horse and drag.
12/ Milling rate 2 tons per day.
13/ No development done.

In calculating net returns royalties paid to the owners of the ground and losses incurred by milling and smelting must be considered. Royalties range from 10 to 40 percent of net returns from the smelter or mill. Mill losses range from as much as 40 percent in amalgamator plants to a minimum of 2 percent in some cyanide mills. Where a concentrate is smelted, losses both at the mill and smelter must be considered. Smelter companies in 1933 paid intermittent shippers for only about 90 percent of the gold in the ore or concentrate.

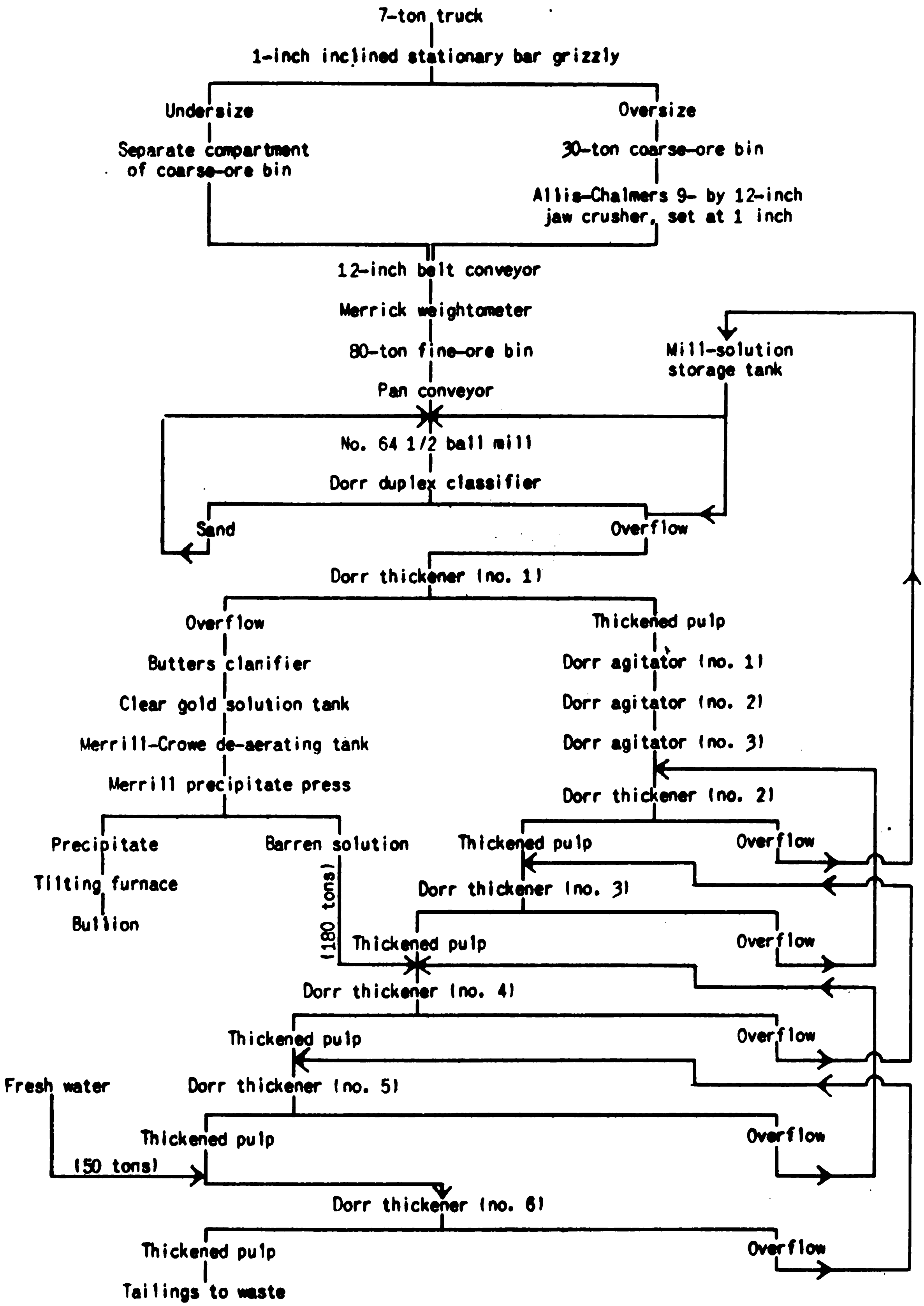

Fig. 14.- Flow sheet of Big Jim (Tellunide) mill.

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